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# MINING METHODS AND COSTS OF THE LAVA CAP GOLD MINING CORPORATION, NEVADA CITY, CALIF.

BY

JOHN W. CHANDLER

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# MINING METHODS AND COSTS OF THE LAVA CAP GOLD MINING CORPORATION, NEVADA CITY, CALIF.[1]

By John W. Chandler[2]

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2/ General superintendent, Lava Cap Gold Mining Corporation, and one of the consulting engineers, Bureau of Mines.

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## INTRODUCTION

The property of the Lava Cap Gold Mining Corporation is in the Nevada City-Grass Valley mining district about 5 miles east of Grass Valley, Calif. At present (July 1940) the plant is mining and milling about 390 tons of gold-silver ore a day; about 310 men are employed.

This paper describes the mining methods used and presents mining costs for 1939, when 60,704 tons were mined by cut-and-fill stoping and 55,676 tons by square-setting.

## ACKNOWLEDGMENTS

The author acknowledges the valuable assistance given him by Otto E. Schiffner, vice-president and general manager, John F. Siegfried, chief engineer, Harry Morton, accountant, and others of the Lava Cap staff in the preparation of this paper.

## HISTORY

Two old mines, the Banner and Central (fig. 1), are embraced in the present holdings of the company. The Banner mine was located in March 1860 and was operated intermittently on a small scale during the next 5 years by various groups. The main Banner ore shoot was discovered in 1865, and the mine shipped and milled about 20,000 tons of ore assaying between $20 and $30 a ton. This ore shoot became lean below the 600 level. Subsequent work did not open up any further ore, so the mine was eventually shut down to remain inactive for 26 years. The Lava Cap Gold Mining Co. unwatered the mine and opened the 850-foot Belshaw shaft in the summer of 1933.

The Central mine was located in 1861. The ledge assayed high in silver, and one of the claims was worked principally for this metal. The other claims were worked for gold. The mine was operated at irregular intervals during the next 57 years. Milling the ore was a serious problem for those early-day miners, who mainly used amalgamation. Tests run as late as 1917 showed extractions as low as 30 percent owing to the large amount of base sulfides in the ore. Introduction of the flotation process made these base sulfide ores amenable to treatment and made it possible to recover more than 95 percent of the gold and silver. After 1918 no important work was done



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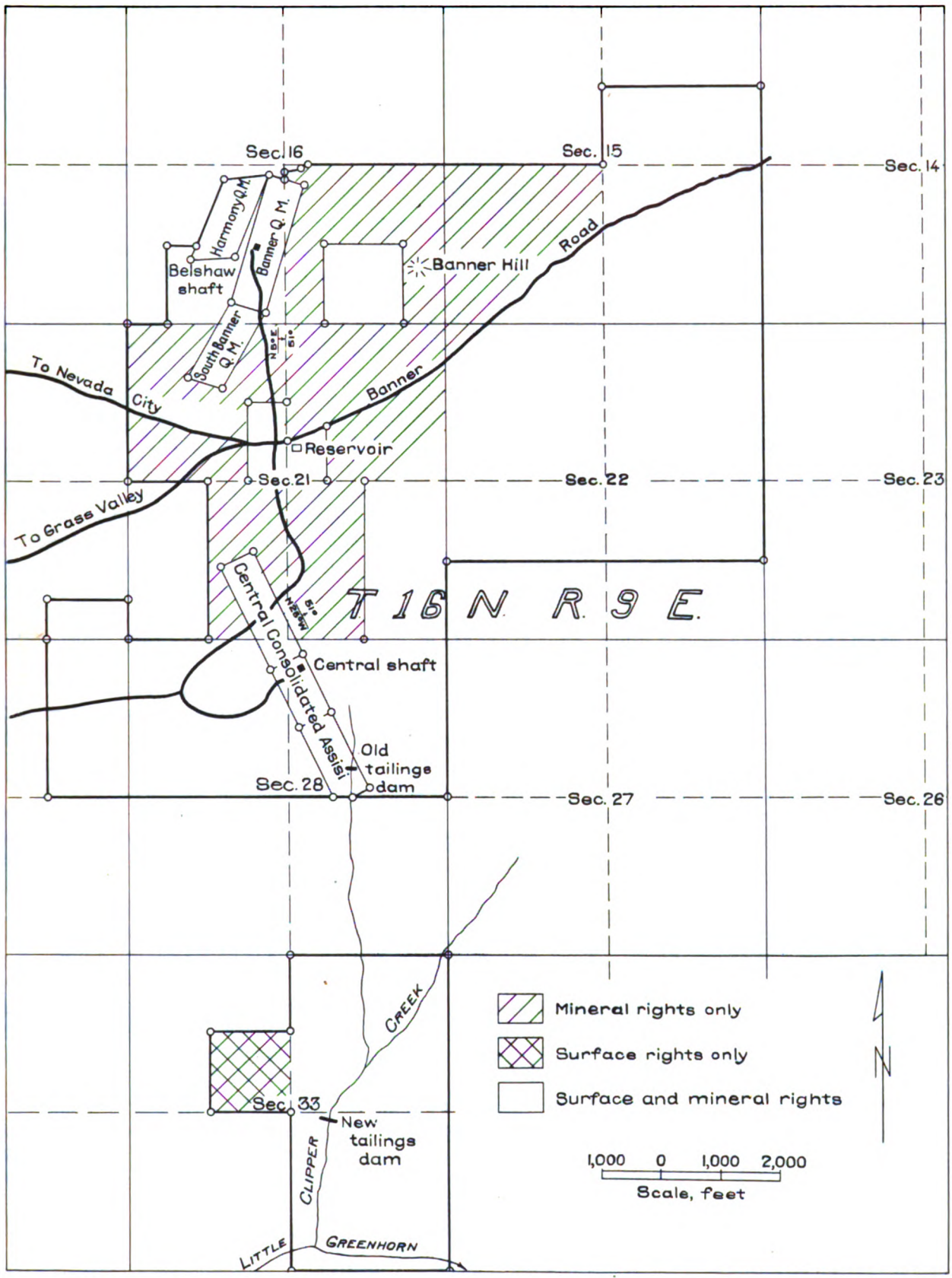


Figure 1.—Location map of Lava Cap Gold Mining Corporation.

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until the present company commenced operations in 1933. An earlier paper has described the status of the mine and mill at the end of the first year's work.3/

## GEOLOGY

The Banner and Central quartz veins are on reverse fault fissures. The Banner vein strikes about due north and dips 50° E., while the Central vein strikes about N. 25° W. and dips 51° NE. About 2,500 feet north of the Central shaft the Central vein joins the Banner vein; it is apparently a split of that structure, as no trace of it could be found to the northwest. The Banner vein continues southward into the footwall of the Central vein. The two veins, for all practical purposes, may belong to but one period of movement and subsequent mineralization, although the Central vein is much smaller than the Banner. This vein has been drifted on for 8,000 feet.

The enclosing walls are argillite, which becomes very siliceous at the Banner end, where it is close to the contact with a mass of granodiorite on the north. Lindgren4/ classifies this argillite as the Calaveras formation of Carboniferous age.

An igneous rock, provisionally called latite, occurs in dikes in the central or southern end of the mine. These do not seem related to the ore shoots. They are older than the vein and serve as useful guides when drifting is done on very weak fractures, as they are always cut by the vein. In the Banner, diorite and andesite dikes have been encountered, which likewise are older than the vein and are unrelated to the occurrence of ore shoots.

A lava flow caps the surface on the high ground between the Banner and Central to a depth of 100 feet or more. At each end erosion exposed the older series of rocks and made possible the discovery of the veins.

Two main ore zones are recognized, one partly under the capping at the Banner and the other making the original Central mine. Smaller ore bodies are found outside these two main zones, but these have not yielded large tonnages of ore.

In the Banner vein widths of 25 feet or greater are common. The average width is about 7 feet. Ore occurs in quartz lenses, as a mass of mineralized quartz stringers within the vein filling, or both. Much movement has taken place both before and after the introduction of the quartz. The main walls have been weakened by fracturing, so large slabs form on the hanging wall. This movement has sheared the vein filling, also, causing it to crack greatly when mined and necessitating heavily timbered and closely filled stopes.

The ore shoots in the Banner ore zone tend to rake flatly to the north on their downward extension and are roughly equal in their dimensions, measured along the strike and dip. As exposed in drifting, many ore shoots have a length of 300 to 400 feet. Although samples cut along the drift give extremely spotty results, a good average grade of ore can be maintained by careful, selective mining.

3/ Chandler, John W., The Lava Cap Enterprise: Eng. and Min. Jour., vol. 135, August 1934, pp. 362-365.

4/ Lindgren, W., The Gold-quartz veins of Nevada City and Grass Valley Districts, California: Geol. Survey 17th An. Rept., pt. 2, 1896, pp. 1-262.



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The large horses of waste within the vein tend to split it considerably. Where these vein splits unite at the ends of the horses their tendency is to form large bodies of quartz, which more often than not result in ore. Some of the highest grade ore in the mine is found at these junctions.

In places the Central vein attains a width of 15 feet or more, but it averages only about 3 feet. There are numerous horses of country rock in the vein, some of considerable size, which in some places split the vein into several branches. When the vein is split into more than two branches by these horses it is rarely minable, as the valuable metals are divided among the various members. In some places one branch of these splits is rich enough to be minable. Where they form two branches both may be minable, but more often only one is profitable.

Numerous rather flat cross stringers of vein material are found where branches of the vein are close together. Following these to the other wall of the vein or to other splits very often discloses additional ore.

Ore shoots at the Central are not as large as at the Banner; they are longer on the dip than on the strike and no rake is noticeable. In general it is found that there is a greater body of minable ore near the ends of horses where vein splits unite than at other places along the vein. Some of the best stopes in the mine are at such points.

The walls in the Central are stronger than those at the Banner, requiring somewhat less timbering, but filling is necessary.

Free gold is finely disseminated through the quartz and sulfides, the latter comprising pyrite, arsenopyrite, sphalerite, and galena. The sulfides within the quartz carry most of the gold. Conversely, much barren pyrite and pyrrhotite are found in the wall rocks.

Silver occurs as argentite always associated with sphalerite. The ratio of gold to silver varies greatly; some ore assayed as high as 500 ounces of silver to the ton.

In general there is so much waste material within the vein walls themselves that selective mining and close sorting are essential to maintain the grade of ore to the mill. The sorted waste has been used in filling the stopes.

## METHODS OF PROSPECTING AND EXPLORATION

At the Banner, where the vein is wide, drifting usually is done on whatever ore is disclosed when the first crosscut is run from the shaft. In the absence of ore a drift is run on the best-looking part of the structure. Every 100 feet crosscuts are run to the foot and hanging walls. The course of the drift is set to follow commercial ore wherever possible. Holes are drilled into the walls every 10 feet while drifting is in progress. The sludges of these holes are assayed and often disclose ore that might otherwise be overlooked.



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At the Central, where the ledge is narrower, both walls usually can be kept within a heading. Drill holes also are used in the Central for sludge-sampling the walls.

When drifting along the ore zone is well under way raises are run at the most favorable points. Sample cuts in these raises often reveal ore valuable enough to justify intermediate drifts; these are found to be one of the major sources of new ore discoveries.

As stoping progresses intermediates are run at the ends of the stopes from the working floors as they are justified, either by assays or by geologic conditions favorable for ore. The waste from such exploration workings usually can be utilized for fill within the stopes themselves.

Crosscutting is done to explore for vein splits. Long crosscuts are being driven at certain horizons to pick up additional structures. Diamond drilling has been used to check the foot and hanging walls for structure.

## METHODS OF SAMPLING AND ESTIMATION OF TONNAGES AND VALUES

Sampling has two main functions: (1) To con rol the grade of ore sent to the mill and (2) to aid in developing ore shoots.

Under the first heading sampling is done at the faces of the stopes in the course of mining. These samples are taken as desired by the foreman and shift bosses; the assays are helpful in controlling stope advance when in ore of marginal value.

Every stope or development heading in ore is sampled by means of a grab sample taken by trammers from the top of every car sent out. These samples are taken on each shift, and because they represent the ore in transit to the mill they indicate to the mill superintendent the grade of mill feed to be expected. The assay returns are entered by the office staff under the headings of the various working places. From assays and corresponding tonnages average grades for each working place and for the mine are calculated month by month. Individual "box samples" vary considerably, but over a period of a month their average corresponds closely to the mill-feed figure. During 1938 the box samples were within 1-1/2 percent of the mill-feed sample.

The purpose of this sampling, aside from its value in mill operation, is to give the management data on the average grade of ore from each stope. Individual costs are kept on each stope, which show the grade of rock that can be mined profitably. Conditions, and therefore costs, differ in every stope, but by the method described it is possible to mine ore close to the margin of profit.

Second, all development faces are sampled by the foremen or shift bosses after every round. The samples are from channel cuts across the vein. They are tied in to survey points and posted daily by the office staff on sheets representing each heading. From these sheets the average width of vein and



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the average grade of ore can be calculated for any length heading desired. These assays, as plotted on plans and sections, form the basis for opening new stopes and often indicate further development work.

No attempt is made to "block out" ore in the strict sense of the term because the ore shoots are too erratic. Ore reserves are calculated in terms of "probable ore." This is defined as the normal expectancy in tons of minable ore, the grade and tonnage being based upon the assay plans and sections and upon the judgment of the engineering department.

## METHOD OF DEVELOPMENT AND MINING

The Banner mine was opened by unwatering the Belshaw shaft, an inclined shaft following the vein to the 800 level. About 600 feet of drifts had been driven on the vein by former operators. This drifting was continued south by the present operators for 2,500 feet and opened two new ore shoots. The mine workings are shown by figure 2.

The Central shaft is a 51° incline in the footwall of the vein and is down 12 levels to a slope depth of about 2,300 feet (fig. 3). When the present company took charge the bottom of the old workings was at the third level. Levels have been established at 200-foot intervals along the incline, crosscuts have been run to the vein, and the vein has been explored by drifting in both directions.

An adit or tunnel has been driven from the Central 5,000 feet to a point over the southern ore shoot at the Banner. This serves as a main haulageway for all ore going to the mill from the Banner mine. This tunnel is the first level at the Central and the 600 level at the Banner, owing to the difference in elevation of the shaft collars. At a point near this southern ore shoot another underground inclined shaft has been sunk in the footwall of the vein, opening two more levels below the 800 level. These have been crosscut to the vein, on which drifts have been run.

Above the Banner tunnel, or 600 level, the 200-foot and 400-foot levels have been established; stoping is proceeding on the 400 level.

Ore from below the tunnel or 600 level of the Banner is hoisted through the underground Banner shaft and hauled by battery locomotive to the Central shaft, where it is transferred through pockets and hoisted through the Central shaft to the mill ore bins at the collar.

Drifts are driven along the vein and raises are started at places most favorable for ore. These raises open ore shoots between levels. They also serve as second exits and form part of the ventilation system of the mine.

## SHAFTS

Both present shafts are 51° inclines driven in the footwall. They have three compartments, two skipways 4 by 5 feet, and a manway 3 by 5 feet.



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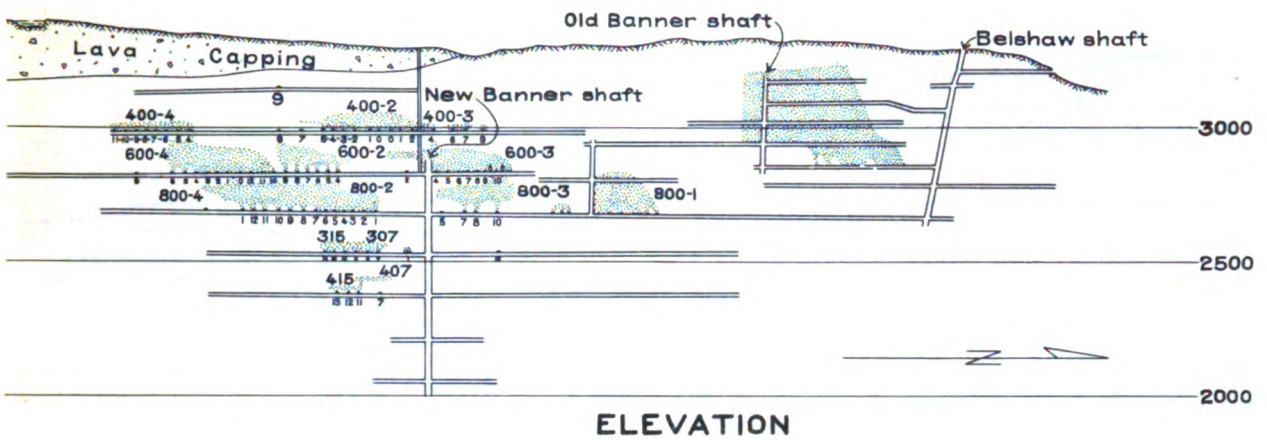


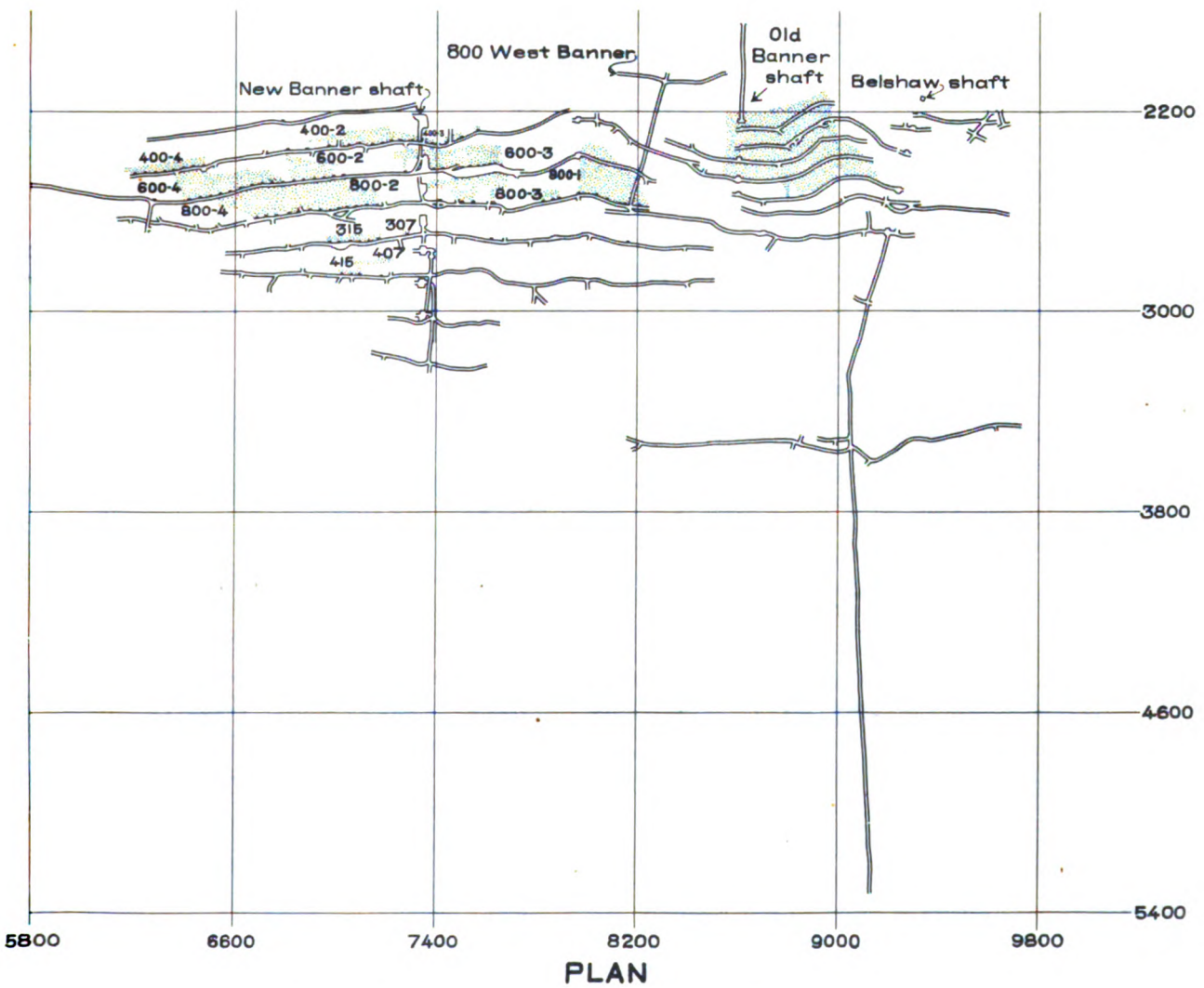


Figure 2.—Underground workings of Banner mine.

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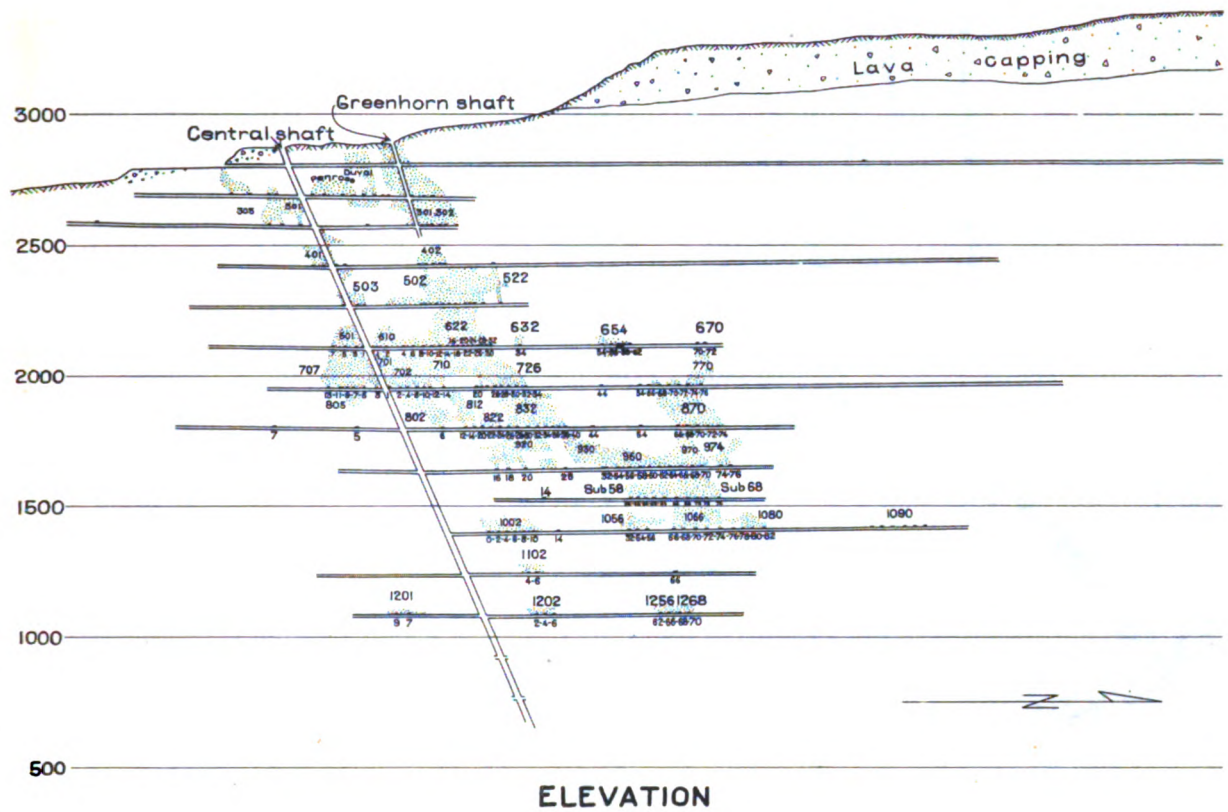


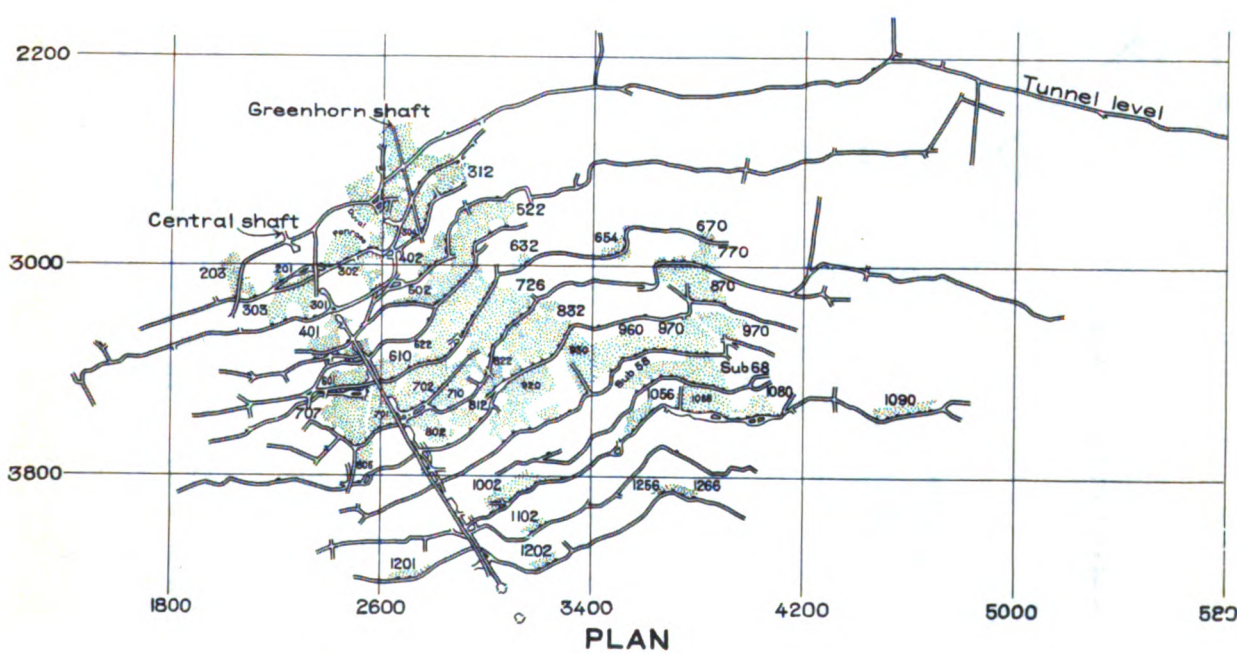


Figure 3.—Underground workings of Central mine.

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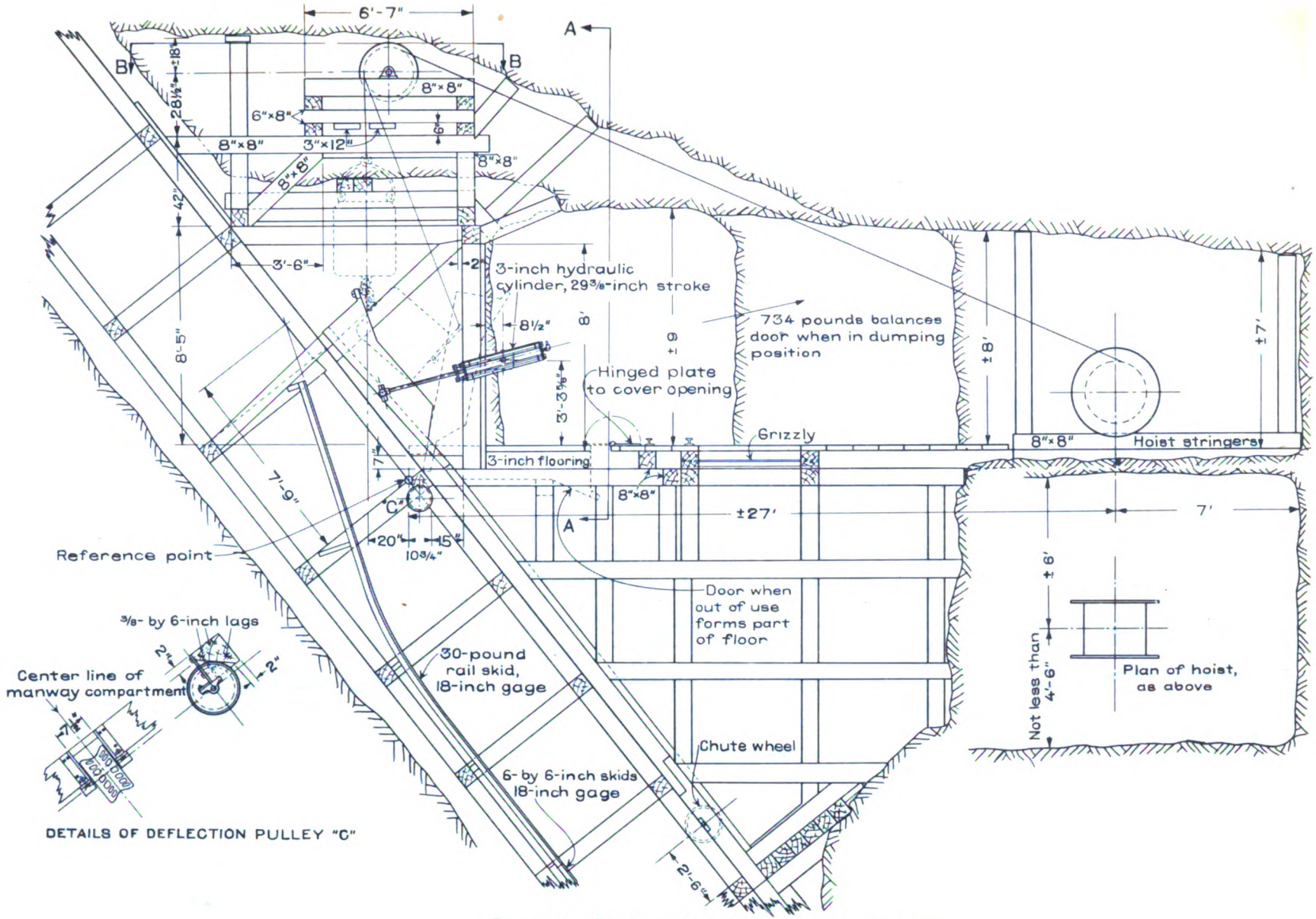


Figure 4.—Shaft-sinking bucket; dump lay-out.

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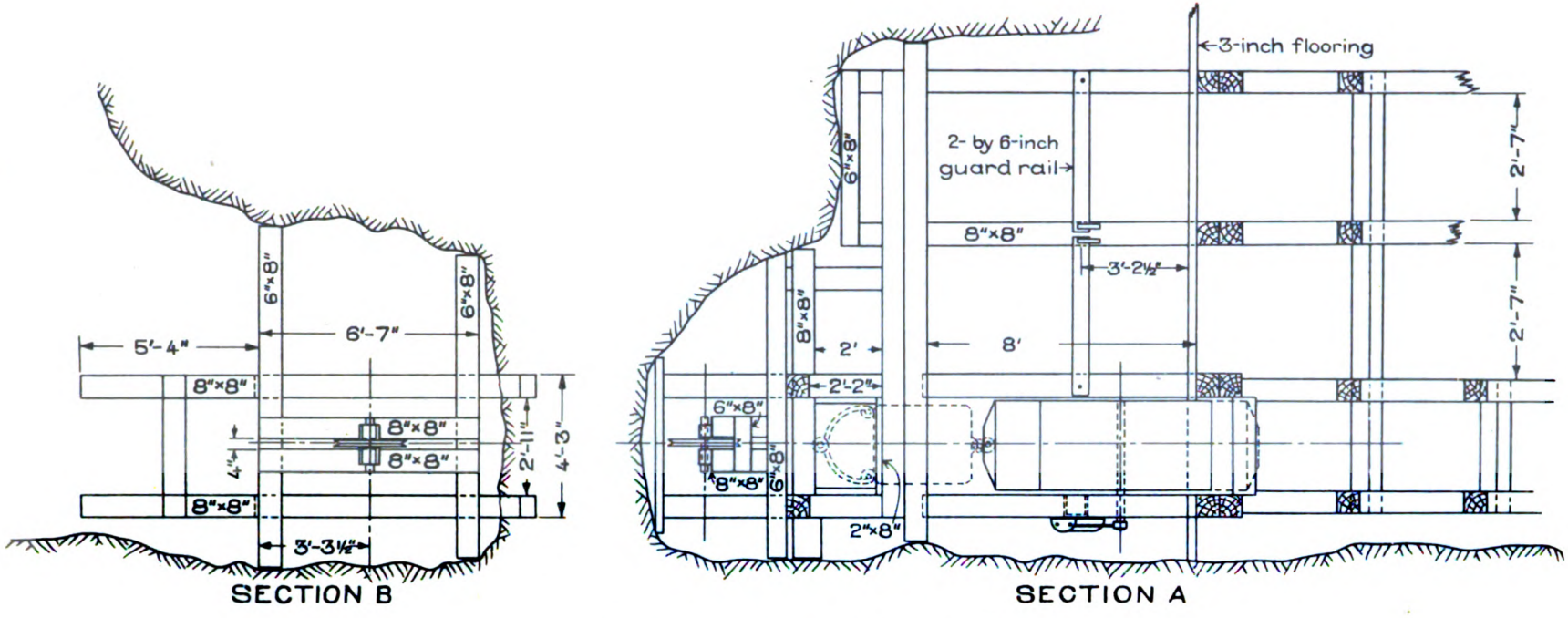


Figure 5.—Shaft-sinking bucket, dump lay-out.

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Shaft-sinking is done from the lowest level with 1,000-pound buckets on skids and a Hendry-Bolthoff air hoist. Hoisting is done through the manway compartment, the skip compartments being protected with two sets of solid rock just below the skip pockets. This practice has been found entirely safe and does not interfere with the normal hoisting of ore from the level. Below the pentice the skidways are angled across the shaft and carried down through the middle compartment.

The sinking buckets are dumped with the conventional tail-chain, trap-door system, the waste being dumped directly into the station pocket for transfer to surface with the skip. This practice eliminates transferring by car. (See figs. 4 and 5).

In the latest construction Port Orford cedar has been used in all wall-plates, stringers, blocking, and wedges. This cedar is extremely resistant to rot and is used untreated. As they are so easily changed, the dividers and posts are of Oregon fir with ends creosoted. Eight by 8-inch wallplates and posts with 6- by 8-inch dividers are used. The shafts are in good ground, and virtually no lagging is necessary.

A cycle of shaft-sinking operations is as follows: Two shifts of four miners each work in the bottom and are serviced by a sinking hoistman. Drilling and blasting are done entirely by one shift and mucking and timbering by another. Blasting is electric. Relatively little water is encountered, and what comes in is handled by a high-lift, "sponge-type" pump. The drilling crew picks and cleans up the bottom, drills a 36-hole round to a depth of 6 feet, and blasts. The other shift comes on 4 hours after blasting time, hangs and blocks a set of timbers, and mucks an average of 110 10-cubic-foot buckets a shift. For a month at a time the crews will average just about 5 feet, or a set, a day. Gardner-Denver D-79 light drifters are used as sinkers with two men to a machine. A standard V-cut shaft round is used and is blasted with 45-percent-strength semigelatin powder. Mucking is by hand. One-inch, quarter-octagon, high-carbon drill steel is used, with the shank end upset to 1-1/4-inch round and lugged. The bits are conventional, forged cross bits.

Pockets are cut while sinking and are timbered integrally with the shaft. They are divided between the two skipways for ore and waste. Skips are loaded through rack and pinion gates with small swinging lips.

## DRIFTS AND CROSSCUTS

Drifts and crosscuts average about 5 by 7 feet in cross section. Drilling and blasting are done on one shift and mucking on the other. Gardner-Denver D-79 and D-89 drifters are in use. The drill steel employed is 1-1/4-inch, round, high-carbon steel with lugged shanks and forged cross bits. Pyramid-cut rounds break best. A 16- or 17-hole round loaded with 110 cartridges of 45-percent-strength, semigelatin powder breaks an average of about 4-1/2 feet.

Spoil is removed chiefly by hand, but in some of the drifting Eimco-Finlay Model 11 loading machines are used; and where the distance from the



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shaft is great battery-locomotive haulage is found economical. Where power loaders are used longer drift rounds are drilled. The use of burnt cuts has increased the length of round considerably, and the extra spoil is handled easily with the loaders.

It is necessary to maintain a switch within 300 feet of the face in the longer headings. A "cherry picker" has been employed successfully; it can pick up a loaded car and swing it to one side, allowing the motor and empties to go by. By means of this device a whole trainload of cars can be loaded out before the motor makes a trip to the dump. The cherry picker can be set up in a space 5 feet wide, 10 feet long, and 8 feet high cut in the side of the heading. It saves the use of switches and sideswiping for siding track. (See fig. 6.)

## RAISES

Raises generally are two-compartment, 5 by 10 feet in size, and stulled in the center, with the chute side lined. Both end-cut and V-cut rounds are used. Two men on a shift form a raise crew, and as gas does not dissipate quickly the raises usually are run on one shift only. The cycle of work after timbering depends chiefly on ground conditions, but ordinarily three or four rounds are shot before timbering is resumed and the bulkhead moved. Chicago-Pneumatic CP-43 stopers are standard equipment for raising.

## MINING (STOPING)

Three stoping methods are followed. In the order of their importance they are: (1) Cut-and-fill; (2) square-sets with fill; (3) open stopes with pillar support.

Most of the ore is mined by the first two methods, with modifications. Various methods have been employed that permit the maintenance of a good grade of ore by very close sorting. One of the best sorting systems is used in connection with the stringer set shown in figure 7. This set is made of 8- by 8-inch stringers with round posts set at 5-foot 6-inch centers and 3- by 12-inch lagging, doubled when shooting upon it. This is satisfactory where walls are competent to stand open for two floors, but in heavier ground it is replaced by square-sets-and-fill.

Where the vein is narrow and stoping is done from stulls sorting is a problem, because the working floor becomes very crowded. Under these circumstances a modification of the stringer-set system is used. Instead of installing stringer sets on each floor, they are only established every 40 or 50 feet up the stope. Enough footwall is shot out to make working room, and the floors are laid. Stoping progresses upward on stulls. The muckers have room to work on the floor and can sort the ore clean, sending the waste through the floor to the open stull section being gobbed below them.

In places it is necessary to remove some of the lean vein to open the ore. This can be done by removing the flooring where this material is being blasted and allowing it all to go into the gob below without having to handle it. This system has proved satisfactory in some parts of the mine.



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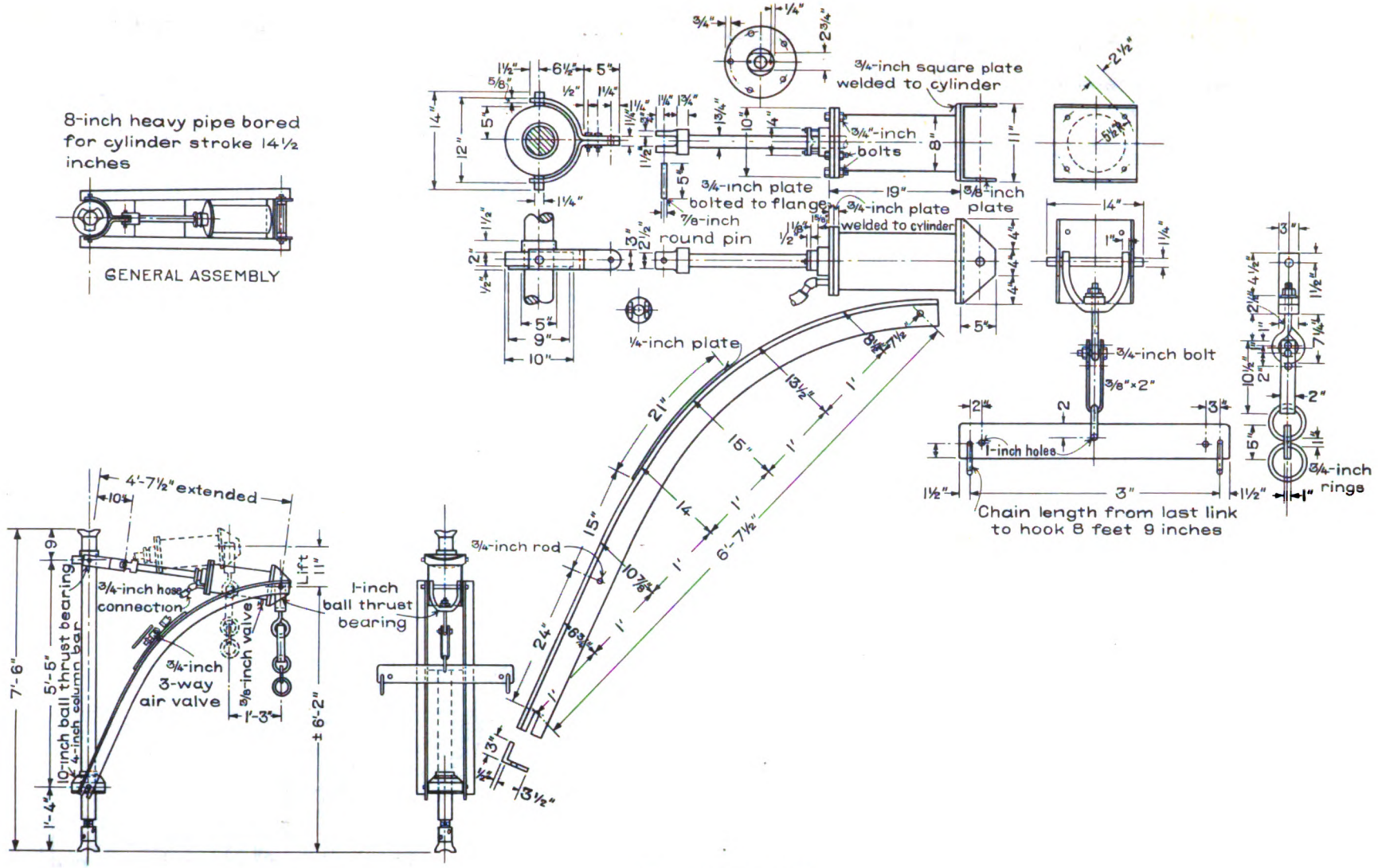


Figure 6.—Car lift.

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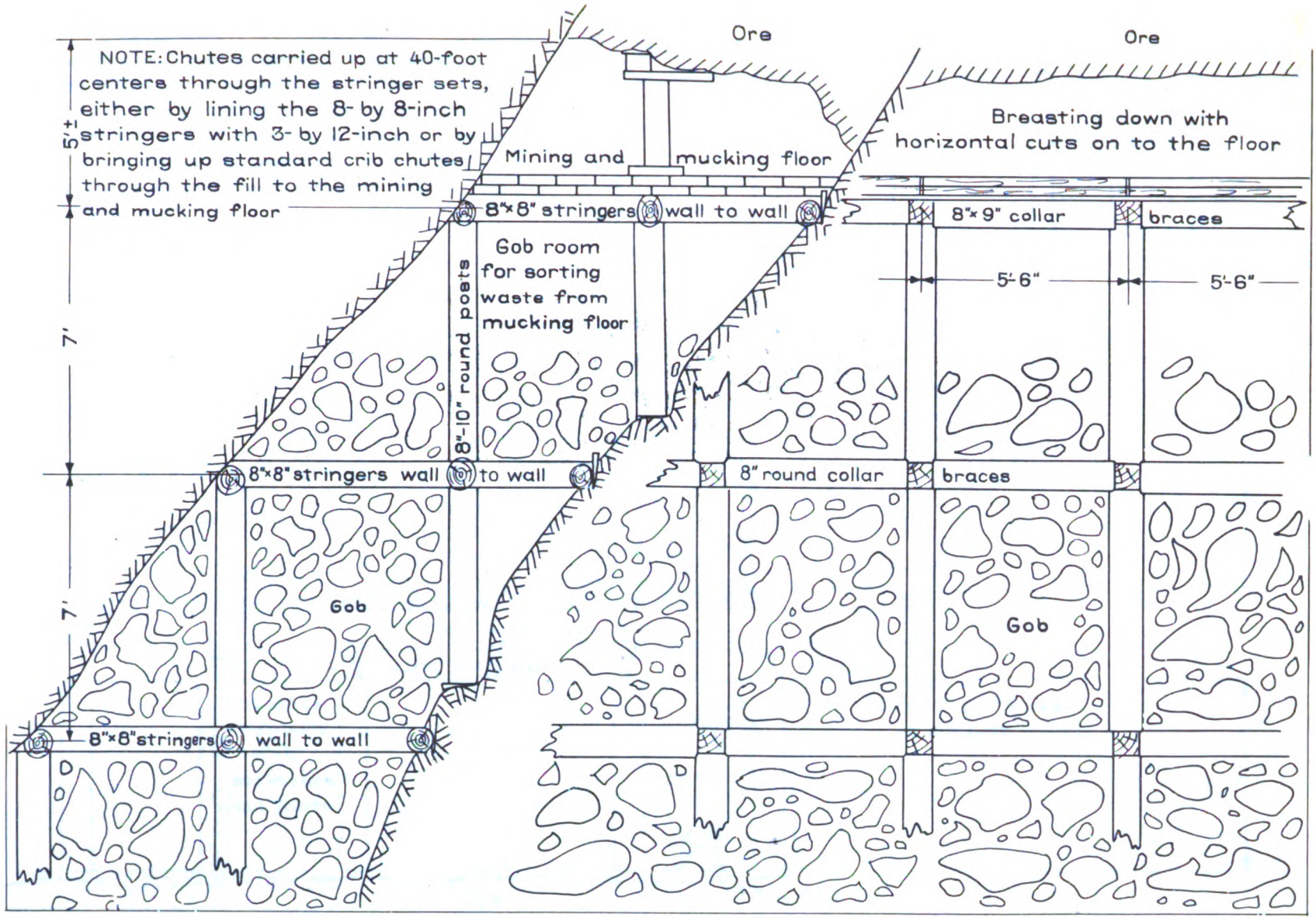
NOTE: Chutes carried up at 40-foot centers through the stringer sets, either by lining the 8- by 8-inch stringers with 3- by 12-inch or by bringing up standard crib chutes through the fill to the mining and mucking floor
5'+
7'
7'
Ore
Ore
Mining and mucking floor
Breasting down with horizontal cuts on to the floor
8"×8" stringers wall to wall
Gob room for sorting waste from mucking floor
8"-10" round posts
8"×9" collar
braces
5'-6"
5'-6"
8"×8" stringers wall to wall
8" round collar
braces
Gob
Gob
8"×8" stringers wall to wall

Figure 7.—Standard 8- by 8-inch stringer sets.

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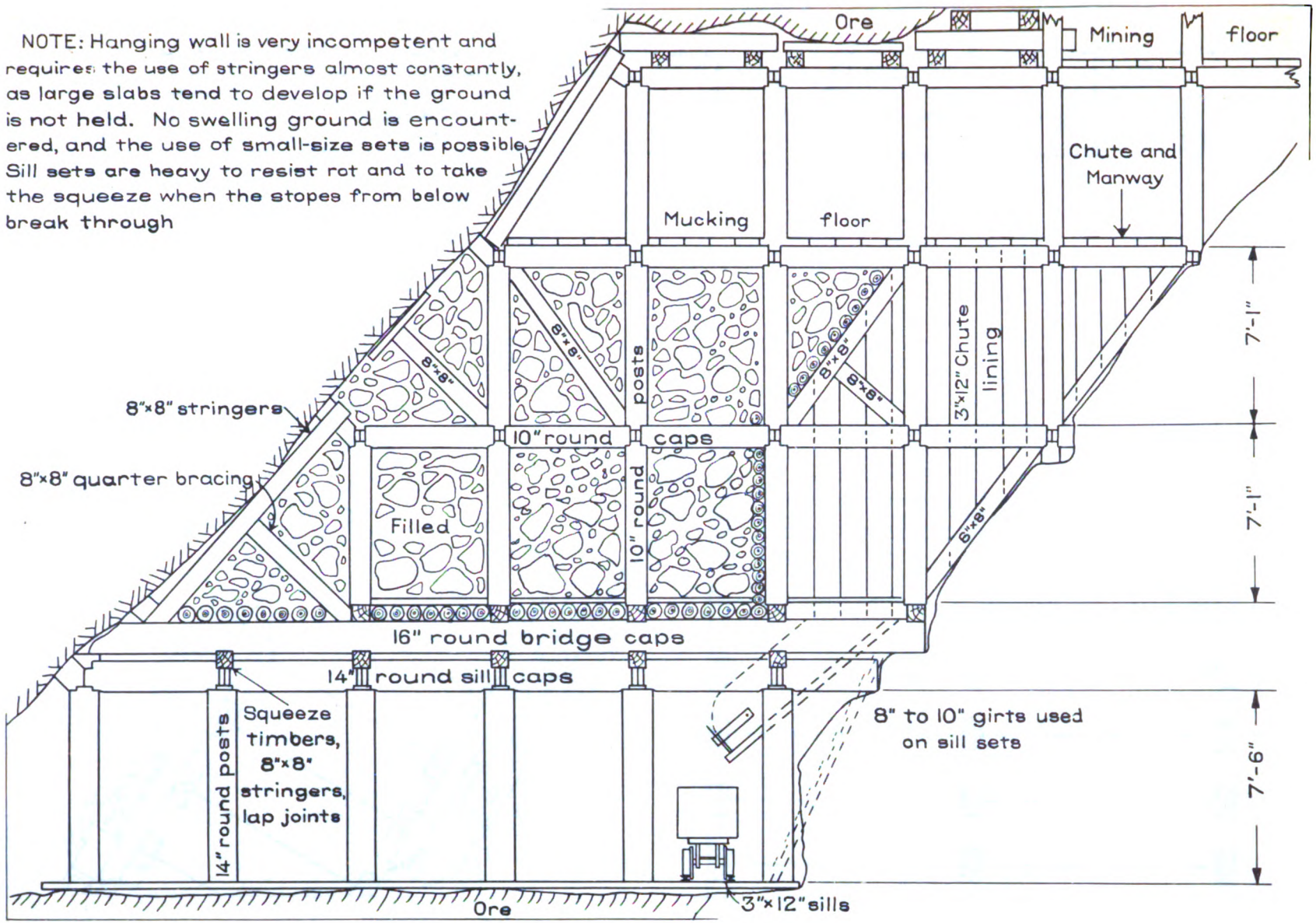


Figure 8.—Typical Banner square-set sill where vein is wide.

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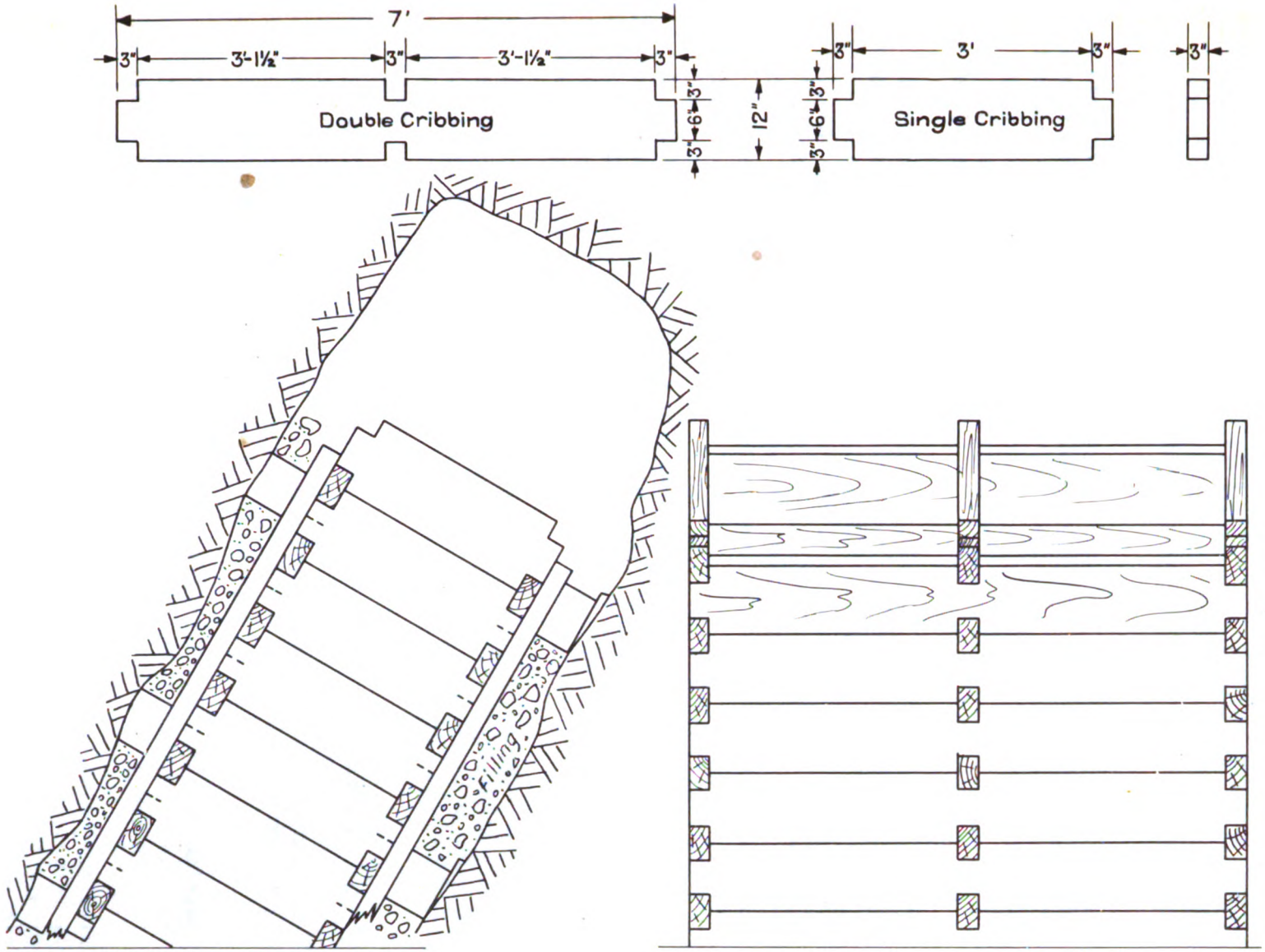


Figure 9.—Single and double 3-by 12-inch standard chute cribbing.

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At the Banner mine the square-set-and-fill method is in use because the badly sheared vein and weak walls must receive immediate support. (See fig. 8.) To a lesser degree stringer sets are used, depending largely on wall conditions. It has been found that as far as wall support is concerned stringer sets are almost worthless and are only strong enough to withstand the shock of blasting and to serve as staging to facilitate the mucking and sorting of the ore.

Open stoping with pillar support is employed chiefly in the narrower lenses of ore in the Central to clean up small sections in the top portion of the mine after most of the ore has been mined. In open stoping it is necessary to mine very clean and avoid dilution. Sorting also is done in these stopes so far as possible. Waste is held in cribs or run into mined-out sections.

All stopes are laid out so that supplies may readily be brought in through raises from the levels by means of steel boats on slides and tugger hoists. Waste from the vein itself is not always sufficient to fill the stopes, and in this event it is necessary to maintain raises and transportation facilities to bring waste into the stopes. Waste-raise passes are alined from level to level for the transfer of waste from development work above to the levels below, requiring stope filling without intermediate tramming.

Light drifters generally are used in stope work where a bench is carried in horizontal cuts. By blasting with flat holes an ore pile can be shot on a floor without scattering, and the waste can be sorted behind the cut without danger of fine ore from the next blast being lost in the waste previously sorted out.

As floor cuts are started from raises, waste is brought in behind the cut in 10-cubic-foot cars running on sectional steel track made of 8-pound rails. In filling some square-set stopes standard concrete buggies serve better than cars, as they can be trundled through a square-set section without laying an elaborate system of tracks.

In small stopes, chutes consist of stulls with gob lagging. In the larger cut and fill stopes chutes are made of stulls from wall to wall for the first 40 feet, thus making pocket room; above that point they are constructed with 3- by 12-inch cribbing. This is readily adaptable; either a single chute or a chute and manway can be cribbed as shown in figure 9.

In the square-sets chutes are carried within the sets themselves. These are lined with 3- by 12-inch plank and gob-lagged on the outside. The tops of all chutes have 8-inch grizzlies made of either timbers or rails.

Various methods of timbering are used in starting stopes from level sills, depending on the conditions of the ground. The simplest consists of stulls with pole sheeting between chutes. Where the foot-wall tends to slough and the hanging wall is fairly good stulls are supported on posts along the footwall side of the track. The foot wall is then lagged and held secure,



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while the post helps support the stull near the hanging wall. Where the ledge is wide sets are stood with caps framed wall to wall and covered with a layer of poles. In the heaviest ground where the main walls are not competent and the width of the vein is great large sill square sets are used with bridge caps, which are sheeted. (See fig. 8.)

## UNDERGROUND TRANSPORTATION

A standard chute set and gate are in use. This was developed because of the previous high cost of installing chutes. It was found that much time was consumed in putting up a chute when men tried to cut and fit the timbers according to the amount of ground shot out. In most instances the chute could have been constructed better if the men could have blasted out a little more room for loading and headroom. If they proceeded to do this, much time was consumed in plugging; therefore a standard chute set has been designed, and before a chute goes in a specified opening is shot out to make room for it. The chute timber is all framed at the surface and fits together readily. The chute sides come intact and have only to be spiked in. The chute gives a trammer room to work. The steel gate is safe to operate by means of a handle, and many accidents common with the old stop-board chutes opened by hand have been eliminated. (See figs. 10A, 10B and 10C.)

Hand tramming is employed on all levels except where ore is transferred from one level to another to eliminate long hand trams, and it is then handled by 1 1/2-ton battery locomotives. Double duty is performed by these motors in returning waste from the shaft to the stopes for filling.

Ore is hauled from the Banner shaft to the Central shaft (5,000 feet) by 4 1/4-ton battery locomotives and 30-cubic-foot rocker-bottom cars in trains of 15 cars to a trip. A man car is shown in figure 11.

In the Central mine 18-cubic-foot end-dump cars are in use both with small locomotives and in hand tramming.

Pockets at the shafts are limited to about 75 tons capacity. The lay-out of ore and waste pockets is shown in figure 12.

Skip dumps are of the Angrove type much used in the Michigan copper country. (See fig. 13.) Overwind trips are set in the dumps to cut the power and apply the brakes if the skip travels too far through them. Electric indicators in the hoist room show whether the dumps are set for ore or waste; these dumps are operated by oil and air cylinders controlled from the hoist-room.

Handling waste for stope fills has always been a problem. There is always plenty of it from new bottom development but not enough in upper levels. At the Central a skip was developed that hooked under the regular skip. Hand wheels control a dumping mechanism that allows the bottom to swing out like the bottom of a power-shovel dipper, deflecting the discharge into the footwall pocket. A hand trip was used to dump the skip because of the chance that an automatic device might cause an accident in the shaft. This solved



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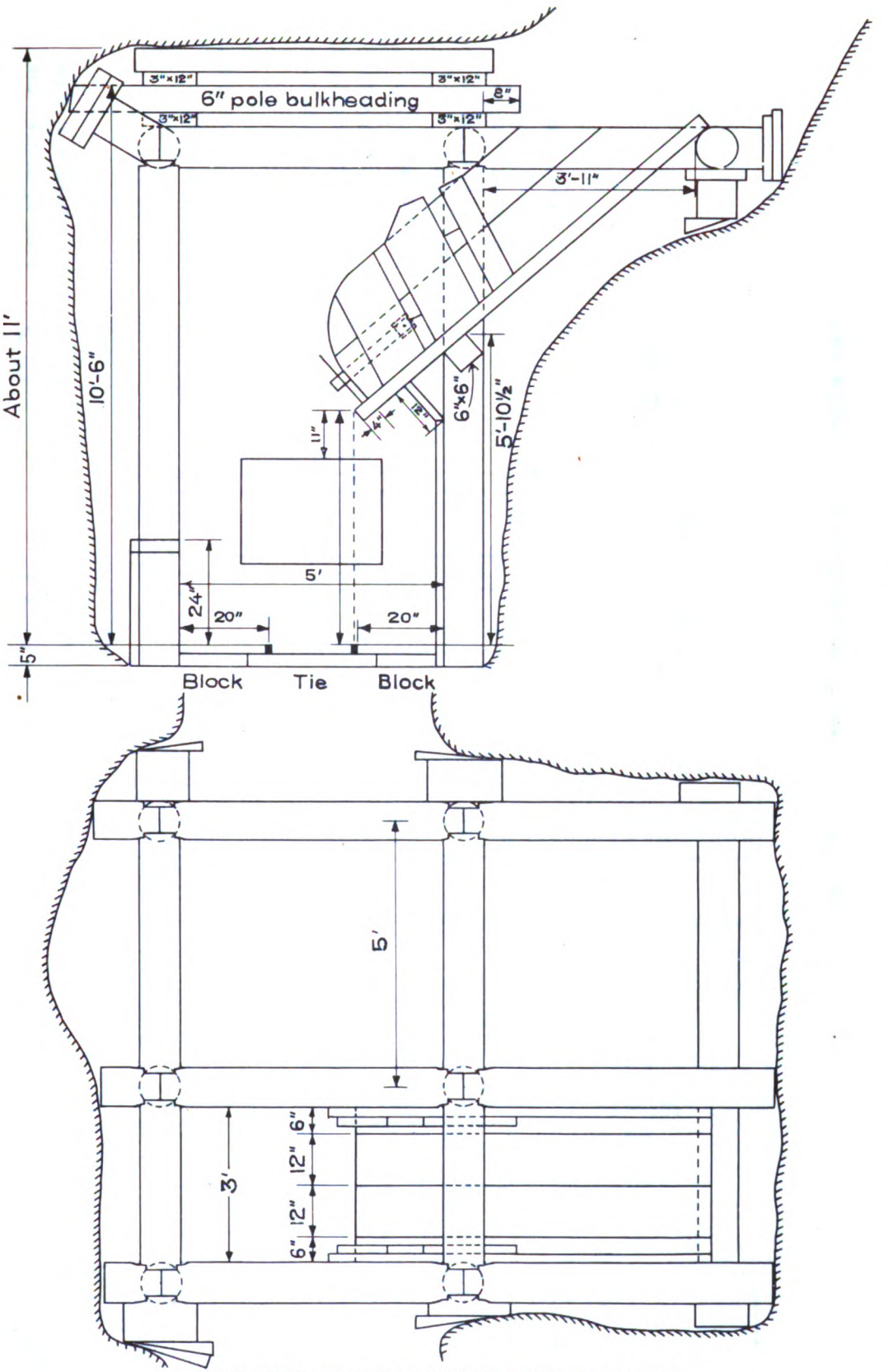


TOP VIEW WITH BULKHEADING OMITTED

Figure 10A.—Assembly of standard drift chute sets.

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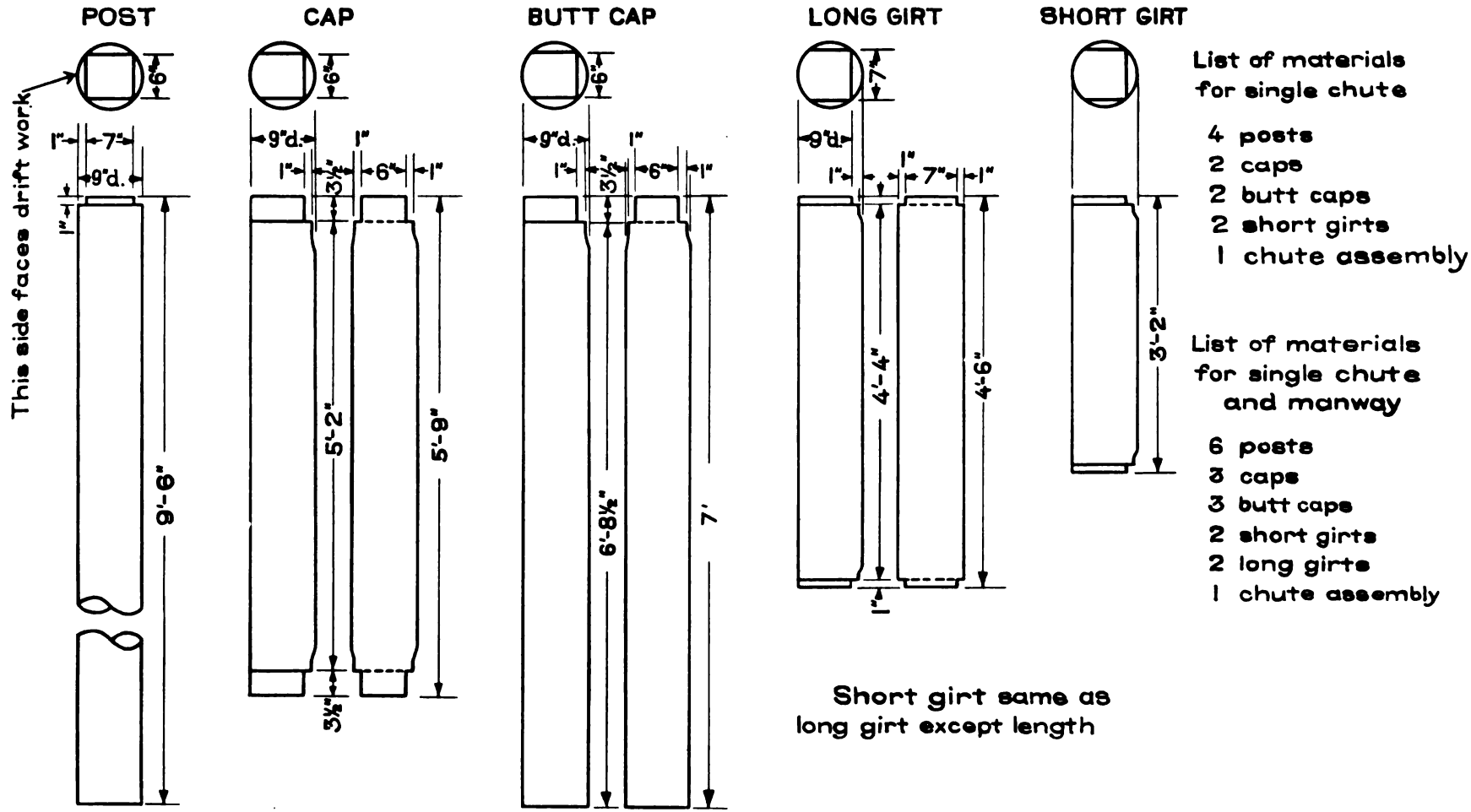


Figure 10B.—Timber framing for drift chute set.

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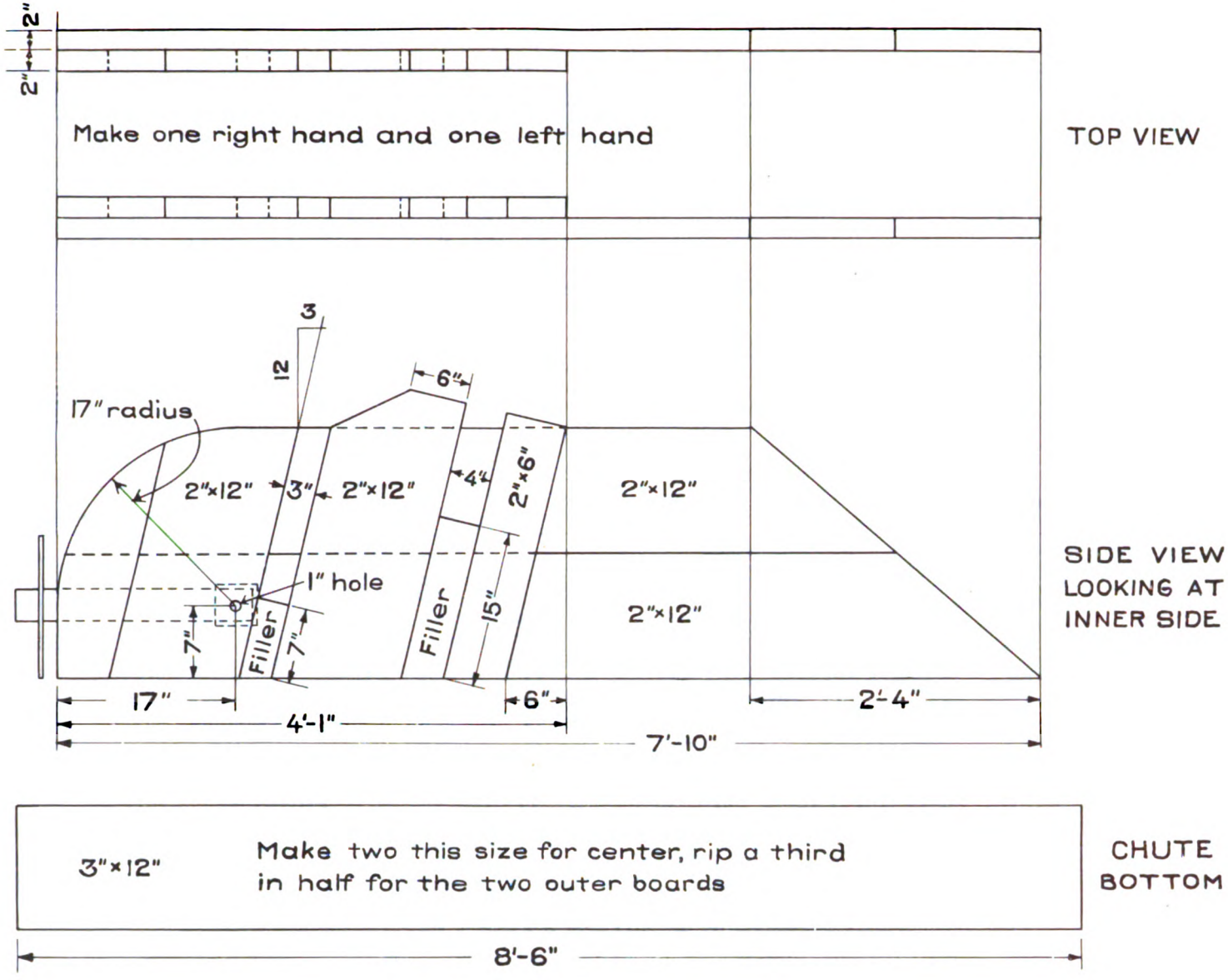


Figure 10C.—Standard chute.

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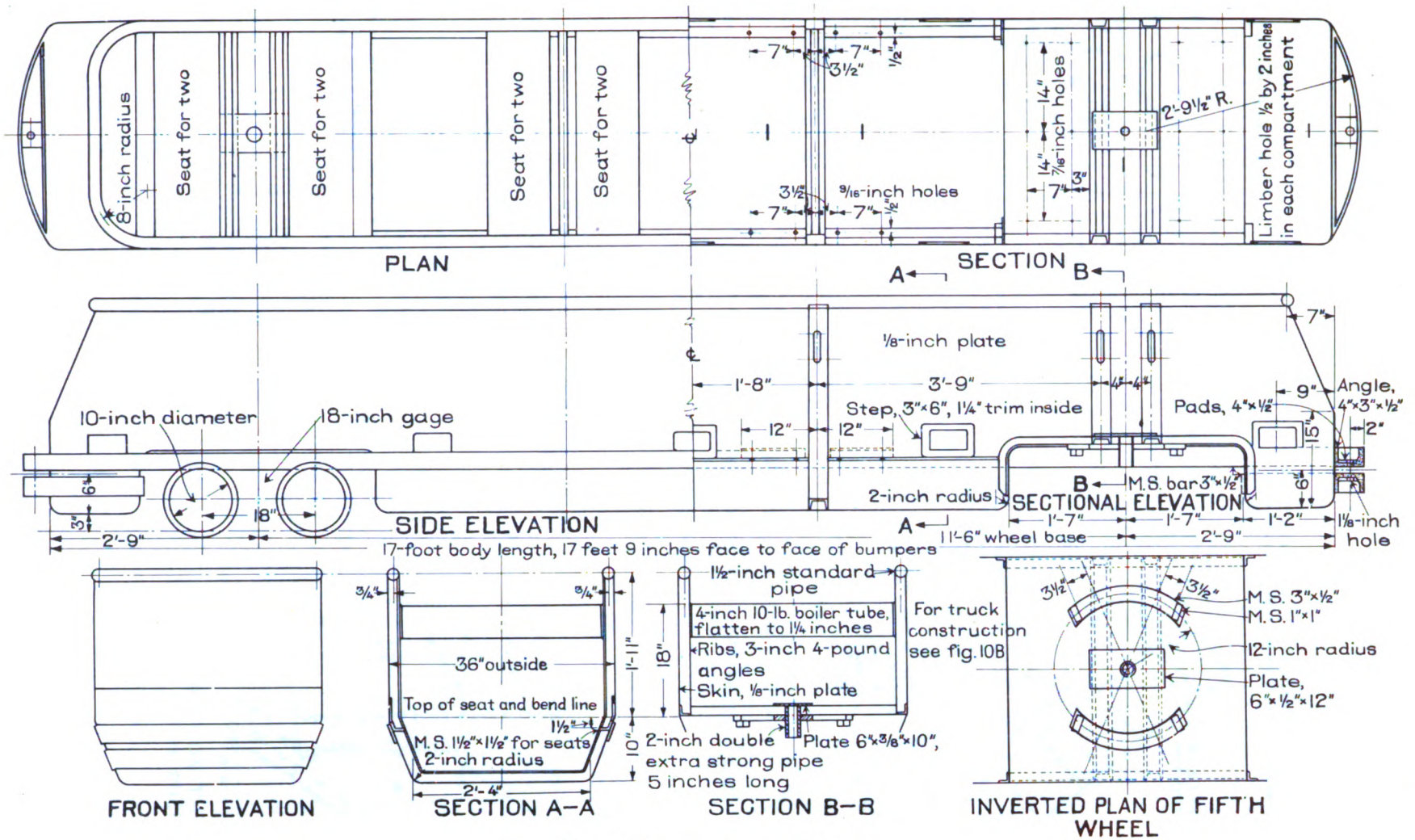


Figure 11.—Tunnel man car.

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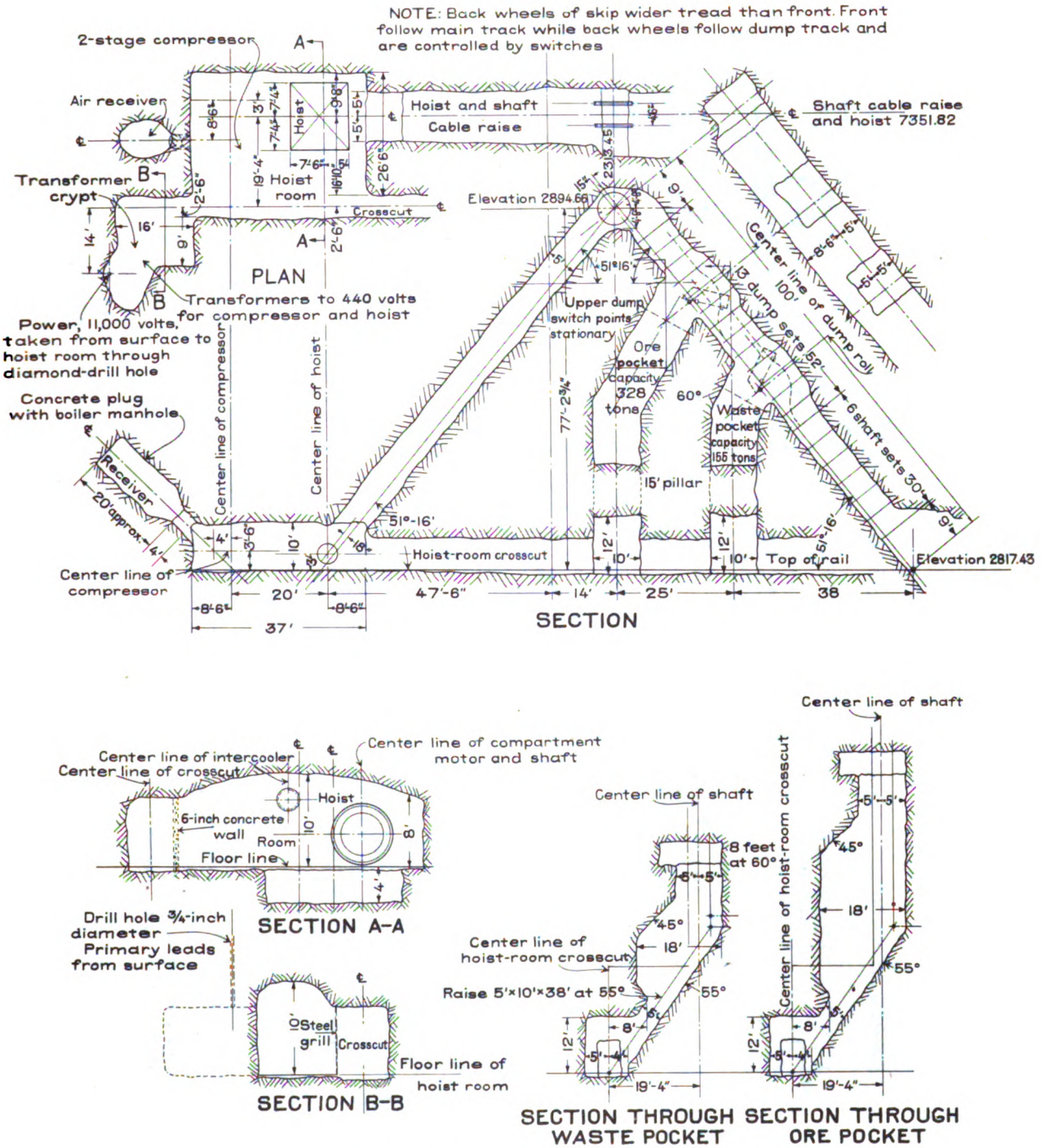


Figure 12.—Rock work of ore and waste pockets.

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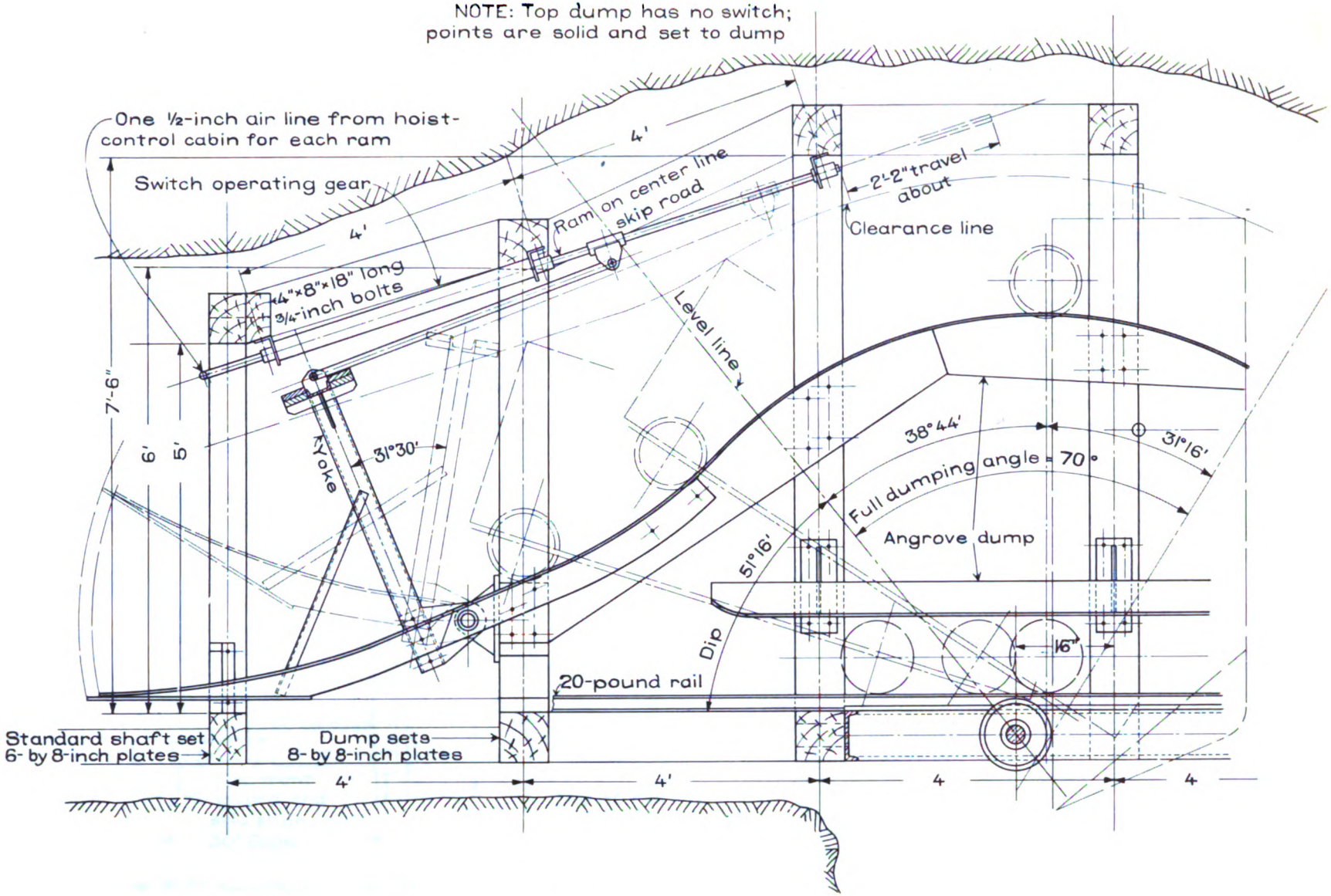


Figure 13.—Installation, switch gear, Angrove dump.

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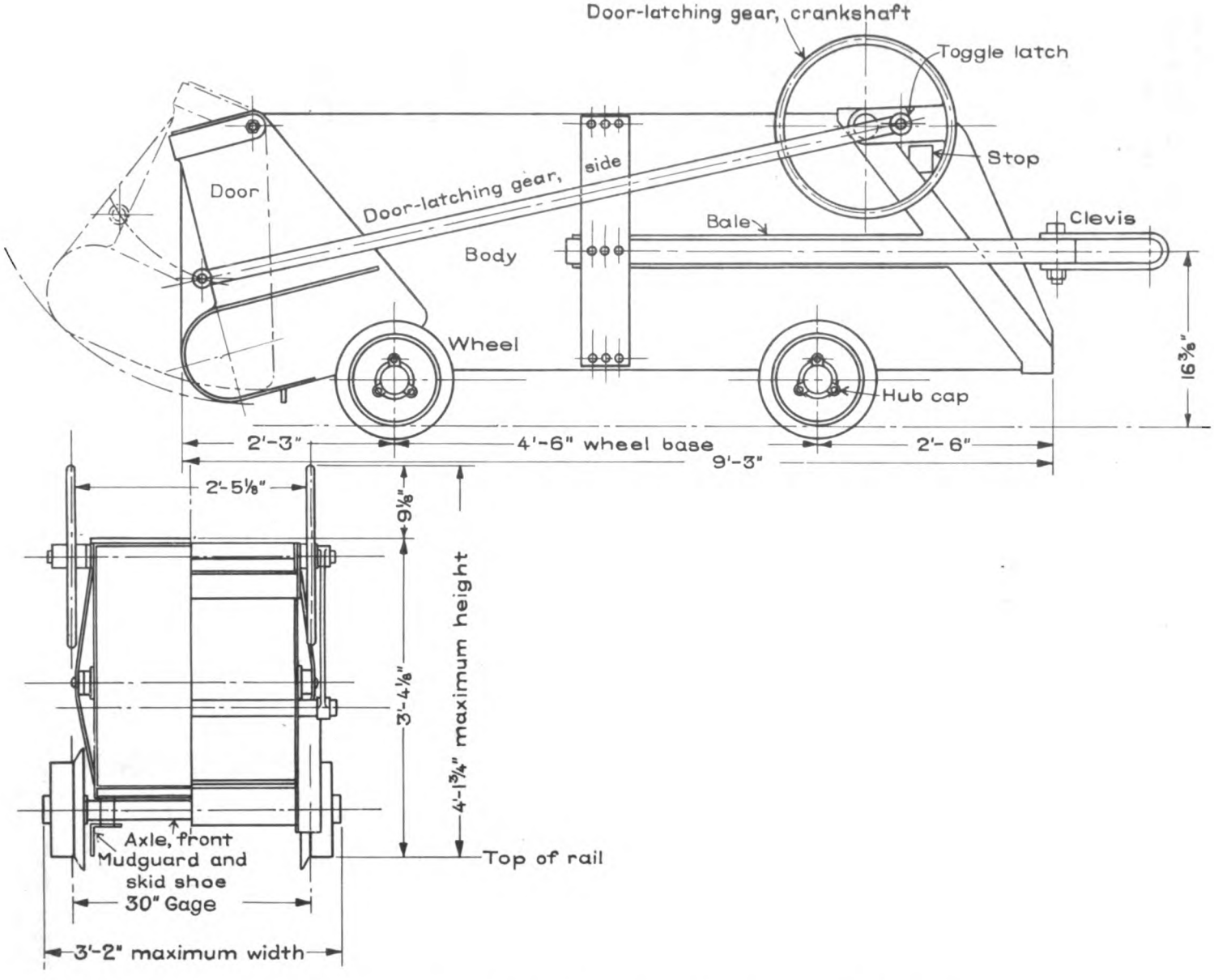


Figure 14.—Waste-transfer skip, 41 cubic feet capacity.

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the caging problem, as it permitted waste to be either hoisted or lowered to a point in the shaft where it could be discharged into a 75-ton storage pocket in the footwall and thence could be hauled on the level for use as stope fill. When not in use the waste skip is stored on the station of an inactive level.

From the level under the waste-storage pockets the waste can be transferred by raises through two or three levels to whichever stope requires it. When the cost of transfer of waste in this fashion becomes too great, another footwall pocket will be cut in the shaft and the same skip used to deliver waste to that point. (See fig. 14.)

Five- by 6-inch guides are used in the Central shaft between each set of rails, and shoes on the skip axles ride the guides in case of derailment. It also prevents a skip from jumping the rails on its up-trip and nosing down under a footwall plate with a wheel, thereby dumping the skip in the shaft.

A Nordberg 300-horsepower hoist with a rope speed of 1,000 feet per minute is used at the Central shaft and an Allis-Chalmers 150-horsepower hoist with a rope speed of 600 feet per minute at the underground Banner. These are equipped with Lilly and Simplex controls, respectively.

## PERCENTAGE EXTRACTION

One-hundred-percent extraction is maintained in stoping work. Dilution by waste is a serious problem, and upon it depends the mill grade. The degree of sorting varies with the grade of rock mined. In some stopes it is done painstakingly, while in others of comparatively low grade it does not pay to sort closely. Sorting is an economic problem different in each stope. In general, 20 percent of all fill used in the stopes is sorted-out material. It has been considered uneconomic to sort waste on the surface.

## WAGE, CONTRACT, AND BONUS SYSTEMS

Wages paid are as follows:

| | |
|---|---|
| Miners and timbermen | $5.84 |
| Muckers | 5.36 |
| Skip tenders | 6.00 |
| Hoistmen | 6.72 |
| Trammers | 5.36 |
| Motormen | 5.84 |
| Switchmen | 5.36 |
| Nippers | 5.84 |

A contract-bonus system is used in most development work. Each heading is paid for at a fixed price per foot, which includes all mining, mucking, and explosives. In raise and shaft work and to a smaller degree in drifts the labor required in timbering is included in the price.

The contract price in any heading is based upon the rate of advance the management expects in that heading. When the first prices were set current



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wages were $5.25 for miners and $4.75 for muckers. Although wage scales have changed from time to time, the original scale is used in computing all contract prices. For example, a drift is being driven with hand mucking where the management would expect a crew on company time to average 3.5 feet per shift with one miner and one mucker on each shift. From experience it is known that this ground will use 25 cartridges of dynamite to the foot advance. A crew on company time will then advance 3.5 feet for the following cost:

| | |
|---|---|
| One miner..... | $5.25 |
| One mucker..... | 4.75 |
| 3.5 feet by 25 equals 87.5 cartridges of explosive | 3.72 |
| | 13.72 |

$\frac{\$13.72}{3.5} = \$3.92$. This is usually raised to an even figure - in this case $4 per foot.

At the end of each month bonus measurements are made by the engineers, and bonuses are computed using the shifts at the base-wage scale. The bonus is converted into the amount per shift over wages at the base scale and is added to wages at the present scale. This obviates the necessity of refiguring contract prices whenever wages change.

The contract sheets are made up and posted on the bulletin board on the 5th of the month, so that the contractors can review them and call any errors or omissions to the attention of the office. Any corrections or errors can then be rectified, and after a day or two the sheets are removed and checks written to cover the bonus earnings.

## VENTILATION

The Central mine is about 1,800 feet deeper than the Banner and is ventilated by means of a 20,000-cubic-foot-per-minute reversible Sturtevant blower situated about 500 feet inside the tunnel between the Banner and Central mines. It is operated as an exhauster under a pressure of 1 inch water gage.

The Central shaft is downcast, and by means of doors and stoppings the air is well-distributed and returns through raises to the tunnel.

The Banner mine has natural ventilation by means of the tunnel.

## PRECAUTIONS AGAINST FIRE

Concrete and steel fire doors are erected at strategic points to guard against fire. Fire buckets and one hundred-gallon drums filled with water



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are placed near all shaft stations and timbered areas for fire protection. These drums are piped with water and are equipped with standard water-hose spuds.

Two mine rescue crews are trained yearly by the Federal Bureau of Mines and receive monthly practice with oxygen breathing apparatus.

The Lava Cap Gold Mining Corporation is a member of the Cooperative Mine Rescue Station in Grass Valley. This station is equipped and maintained by four of the larger mining companies in the district for mutual protection in case of disaster. It is provided with 11 McCaa 2-hour oxygen breathing apparatus, 5 All-Service gas masks, and other equipment.

## FIRST-AID AND SAFETY ORGANIZATION

All men at the Lava Cap are trained yearly in first aid by the Federal Bureau of Mines.

Once a month each shift holds a safety meeting presided over by the safety engineer. Each shift has a safety committee of three men, one man being selected every 2 months by popular vote. The newest member takes his place as the chairman retires, and the oldest man on the committee automatically takes the chair. This insures that the chairman has served 4 months on the committee and consequently should be qualified to handle it. There are four such committees - two at the Banner and two at the Central. Their periodic inspection trips are so timed that the entire mine is fully covered by an inspection every 2 weeks. The committees list suggestions, which they discuss with their respective foremen. The foremen then rectify any undesirable conditions or practices. At the following safety meeting the reports of the committees are read, and the foremen state what action has been taken on each suggestion.

After the reports of safety committees safety suggestions are called for from the floor, and many valuable suggestions have been offered.

An executive inspection is made of the whole mine, both surface and underground, every 60 days by the general superintendent, safety engineer, and both mine foremen, accompanied on the surface inspection by the master mechanic and mill superintendent.

The safety engineer has a splendid opportunity to take up a regular program of safety education with the men and to teach them to think in terms of safety.

## EMPLOYMENT PRACTICES

It has long been recognized that the occurrence of accidents depends partly on the class of workers employed. For this reason the Lava Cap gives particular attention to the selection of its employees. An employment office is conveniently near town. Hiring is done three times a week by the safety



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engineer; every applicant for work is interviewed. After being interviewed each man either is told that he cannot be considered for employment for reasons given or he is asked to return at the next regular hiring day for a definite reply as to his chances for employment. The last request is made to enable the safety engineer to gather a group of likely-looking men together and study their adaptabilities. From this group he can select the best to meet the current need for replacements. When the hiring day arrives he selects the men he wants and lets the rest of them go. The men selected are sent to an examining physician.

When a man returns from the doctor's office with a satisfactory report he is given a "rustling card" good for 3 months. No promises or commitments are made except that any man who has obtained a rustling card will go to work within a reasonably short time. Only enough men to satisfy current needs are given rustling cards. It is found that when this system is adhered to men will continue to seek employment because they know they will not have to wait very long for work.

No men are "led along" waiting for jobs. At times when the turn-over is small it is necessary to tell men that the chances of employment are not very good and ask them to return after a short time. This treatment of men develops confidence between the men and the safety engineer and leads them to do as he asks. Men who are well-fitted for work are given a chance to obtain a card when they return, provided the rustling line is not overcrowded. Preference is given to former employees.

This line of approach has been very successful. It is recognized that this is the best method for close contact with men who are not actually employed by the company. The approach to the men is kept as pleasant as possible, and care is taken to see that men who are refused employment do not leave with ill feeling toward the company.

Men who are working under good conditions tell others. The best men in the mine are acquainted with other good men who attempt to gain employment. By close selection a better-than-average crew is maintained and the work is done efficiently.



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TABLE 1. - Underground costs per ton of ore hoisted during 1939, Central mine, Nevada City, Calif. 1/

| | Labor | Super-vision | Compressed-air drills and steel | Power, cost | Explo-sives | Timber | Other supplies | Total |
|---|---|---|---|---|---|---|---|---|
| evelopment..... | $0.841 | $0.044 | $0.124 | $0.013 | $0.291 | $0.033 | $0.137 | $1.483 |
| ining (stoping).. | 1.801 | .095 | .212 | .034 | .183 | .203 | .264 | 2.792 |
| ransportation (underground).... | .362 | .019 | - | .055 | - | - | .053 | .489 |
| eneral under-ground expense... | .207 | .012 | - | - | - | - | .033 | .252 |
| urface expense (directly applic-able to under-ground)..... | .372 | .023 | - | .008 | - | - | .067 | .470 |
| Total..... | 3.583 | .193 | .336 | .110 | .474 | .236 | .554 | 5.486 |

/ 60,704 tons ore hoisted during year by cut-and-fill method.

TABLE 2. - Underground costs per ton ore hoisted during 1939 at Banner mine, Nevada City, Calif. 1/

| | Labor | Super-vision | Compressed-air drills and steel | Power, cost | Explo-sives | Timber | Other supplies | Total |
|---|---|---|---|---|---|---|---|---|
| evelopment..... | $0.874 | $0.046 | $0.107 | $0.024 | $0.207 | $0.099 | $0.141 | $1.498 |
| ining (stoping)..... | 1.711 | .089 | .162 | .058 | .083 | .308 | .271 | 2.682 |
| ransportation (under-ground)..... | .440 | .024 | - | .056 | - | - | .073 | .593 |
| eneral underground expense..... | .141 | .008 | - | - | - | - | .023 | .172 |
| urface expense (directly applicable underground)..... | .427 | .027 | - | .006 | - | - | .083 | .543 |
| Total..... | 3.593 | .194 | .269 | .144 | .290 | .407 | .591 | 5.488 |

/ 55,676 tons ore hoisted during year by square-set method with filling.



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TABLE 3. - Summary of costs, in units of labor, power, and supplies, for 1939, Central mine, Nevada City, Calif. 1/

| | Development | Mining (stoping) | Total |
|---|---|---|---|
| Labor (man-hours per ton ore hoisted): | | | |
| Breaking (drilling and blasting) | 0.489 | 0.706 | 1.195 |
| Timbering and filling | .060 | .473 | .533 |
| Mucking | .373 | 1.177 | 1.550 |
| Haulage and hoisting | .112 | .331 | .443 |
| Supervision | .073 | .190 | .263 |
| General | .231 | .580 | .811 |
| Total labor underground | 1.338 | 3.457 | 4.795 |
| Average tons per man-shift | 5.726 | 2.354 | 1.668 |
| Labor, percent of total cost | 65.10 | 71.71 | 69.48 |
| Average tons per man-shift on surface properly chargeable to underground operation) | 43.731 | 21.867 | 14.578 |
| Power and supplies: | | | |
| Explosives (pounds per ton ore hoisted) | 1.217 | 1.747 | 2.964 |
| Gelodyn No. 3 (45 percent volume strength) | | | |
| Timber (B.M.per ton ore hoisted) | .791 | 7.120 | 7.911 |
| Poles (linear feet per ton ore hoisted, size 8-16-inch) | .221 | .977 | 1.198 |
| Total power (kw.-hr. per ton ore hoisted) | 11.294 | 25.761 | 37.055 |
| Air compression | 6.314 | 11.472 | 17.786 |
| Hoisting | 2.057 | 7.169 | 9.226 |
| Pumping | 1.725 | 4.204 | 5.929 |
| Ventilation | .356 | .867 | 1.223 |
| Lighting | .302 | .736 | 1.038 |
| Shops (portion chargeable underground) | .421 | 1.024 | 1.445 |
| Battery charging sets (haulage) | .119 | .289 | .408 |
| Other supplies, percent of total supplies and power | 31.09 | 38.65 | 35.65 |
| Supplies and power, percent of total cost | 31.98 | 24.99 | 27.35 |
| Compensation insurance, assaying, percent of total cost | 2.92 | 3.30 | 3.17 |
| Percent of total cost | 33.69 | 66.31 | 100.00 |

1/ 60,704 tons ore hoisted during year by cut-and-fill stoping method.



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TABLE 4. - Summary of costs, in units of labor, power, and supplies, for 1939, Banner mine, Nevada City, Calif. 1/

| | Development | Mining (stoping) | Total |
|---|---|---|---|
| abor (Man-hours per ton ore hoisted): | | | |
| Breaking (drilling and blasting) | 0.416 | 0.446 | 0.862 |
| Timbering and filling | .162 | .669 | .831 |
| Mucking | .272 | 1.116 | 1.388 |
| Haulage and hoisting | .160 | .375 | .535 |
| Supervision | .060 | .156 | .216 |
| General | .212 | .555 | .767 |
| Total labor underground | 1.282 | 3.317 | 4.599 |
| Average tons per man-shift | 6.016 | 2.447 | 1.740 |
| Labor, percent of total cost | 67.12 | 71.09 | 69.67 |
| Average tons per man-shift on surface (properly chargeable to underground operation) | 40.109 | 20.056 | 13.370 |
| ower and supplies: | | | |
| Explosives (pounds per ton ore hoisted) (Gelodyn No. 3, 45 percent volume strength) | 1.259 | .454 | 1.713 |
| Timber (B.M. per ton ore hoisted) | 2.384 | 8.968 | 11.352 |
| Poles (linear feet per ton ore hoisted), 8- to 16-inch | .459 | 2.059 | 2.518 |
| Total power (kw.-hr. per ton ore hoisted) | 7.873 | 12.847 | 20.720 |
| Air compression | 2.592 | 4.245 | 6.837 |
| Hoisting | 2.449 | 1.633 | 4.082 |
| Pumping | 1.868 | 4.597 | 6.465 |
| Ventilation | .186 | .456 | .642 |
| Lighting | .233 | .575 | .808 |
| Shops (portion chargeable underground) | .156 | .383 | .539 |
| Battery-charging sets (haulage) | .389 | .958 | 1.347 |
| Other supplies, percent of total supplies and power | 32.39 | 38.00 | 35.79 |
| Supplies and power, percent of total cost | 30.03 | 25.58 | 27.17 |
| Compensation insurance, assaying, percent of total cost | 2.85 | 3.33 | 3.16 |
| ercent of total cost | 35.53 | 64.47 | 100.00 |

1/ 55,676 tons of ore hoisted during year by square-set with filling method.



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TABLE 5. - Details of costs, in units of labor, power, and supplies, Central and Banner mines, Nevada City, Calif.

Development

| | Sinking | Cross-cutting | Drifting | Raising | Toll, all development |
|---|---|---|---|---|---|
| Size of headings | 7 x 14 | 5 x 7 | 5 x 7 | 5 x 10 | |
| Timbered or not | Regular shaft sets | None | Small amount | Stulls with lining | |
| Physical properties of rock | Hard | Hard | Medium to soft | Medium | |
| Labor (man-hours per foot): | | | | | |
| Breaking (drilling and blasting) | 7.258 | 2.588 | 2.043 | 3.989 | 2.512 |
| Timbering | 3.633 | .093 | .149 | 2.487 | .603 |
| Mucking | 6.311 | 2.388 | 1.933 | .120 | 1.768 |
| Haulage and hoisting | 2.832 | 1.353 | .948 | 1.863 | 1.165 |
| Supervision | .415 | .422 | .339 | .557 | .382 |
| Total labor | 20.449 | 6.844 | 5.412 | 9.016 | 6.430 |
| Average feet per round | 4.72 | 3.73 | 4.26 | 4.55 | 4.27 |
| Power and supplies (per foot): | | | | | |
| Explosives (pounds per foot) | 18.930 | 12.632 | 8.271 | 7.333 | 8.647 |
| Timber (per foot): | | | | | |
| Poles (linear foot) | 0 | 0 | .18 | 2.52 | |
| Sawed (B.M.) | 137.00 | 0 | .63 | 24.87 | |
| Total power (kw.-hr. per foot) | 146.849 | 78.845 | 50.270 | 53.712 | 54.859 |
| Air compression | 87.741 | 27.820 | 24.357 | 22.721 | 25.752 |
| Hoisting | 32.728 | 11.468 | 11.476 | 16.389 | 12.752 |
| Pumping | 12.173 | 31.007 | 8.699 | 8.361 | 10.189 |
| Ventilation | 3.449 | 2.372 | 1.456 | 1.443 | 1.559 |
| Lighting | 3.382 | 2.327 | 1.428 | 1.415 | 1.529 |
| Shops (portion chargeable underground) | 3.758 | 2.584 | 1.586 | 1.572 | 1.669 |
| Haulage (Battery charger sets) | 3.618 | 1.267 | 1.268 | 1.811 | 1.409 |
| Other supplies (cost per foot) | $7.79 | $0.78 | $0.72 | $1.20 | $0.97 |
| Labor, percent of total cost | 66.74 | 63.96 | 63.96 | 64.82 | 64.52 |
| Power and supplies, percent of total cost | 28.36 | 30.99 | 30.45 | 29.62 | 30.02 |
| Compensation insurance, assaying, shops, percent of total cost | 4.90 | 5.05 | 5.59 | 5.56 | 5.46 |



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